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By G. L. Oldright

ERRATUM SHEET

Page 4, Footnote 11, Change to read

11/ Oldright, G. L., Ravitz, S. F., and deRycker, Henry, Some Experiments on Sintering Lead Sulphate Products: Trans., A.I.M.E., vol. 159, 1944, p. 81.

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DEPARTMENT OF THE INTERIOR
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G. L. OLDRIGHT

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UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

RECONVERSION PROBLEMS IN LEAD METALLURGY[1]

G. L. Oldright[2]

The generally accepted forecast respecting the price of domestic lead is that it will remain favorable. Uncertainty is felt, however, as to whether enough ore will be developed by present vigorous exploratory campaigns to permit operators to maintain production on the scale necessary to insure low unit costs. The metallurgist must adopt the simplified flow sheets and mechanization necessary to increase production per man-day. Hence, at widely separated concentrators, discarding of gravity units of small capacity or those requiring too much individual attention has been governed more by the desirable properties of the concentrates than by the particle sizes at which the concentrate minerals were liberated, or by the recovery of the minerals. Perhaps the most significant recent changes in concentration have been made in grinding, classifying, and thickening practices to simplify the flow sheets. Flotation of coarser particles by all-flotation methods has been favored to avoid losses incidental to overgrinding the slime, to lessen the amount of flotation reagents, and to avoid the excessive cost of finer grinding. In order to discard gangue at coarser sizes without choking the flotation circuit with sand, it is necessary to free the valuable minerals by preferential grinding.

At a Broken Hill concentrator using all-flotation, the feed, consisting of a rich lead-zinc-iron sulfide mixture in a heavy, hard, siliceous gangue, is ground to 16 percent plus 65-mesh and only 45 or 50 percent minus 200-mesh by circulating 150 tons to a ball mill per 35 tons of original feed. The siliceous sand does not settle in the flotation cells, as the feed is a pulp of 50 percent solids. The reagents used are measured in thousandths to a tenth of a pound. The cleaned lead concentrate, which contains 76 percent lead, is 78 percent minus 200-mesh material. The zinc concentrate is 48 percent minus 200-mesh material.

J. B. Huttl[3] notes the practice of the American Zinc, Lead, & Smelting Co. of using fine sieving to attain the above objectives. To avoid sliming the relatively coarse galena, the undersize from the last screen of the stage-crushed ore is sent directly to a ball-mill classifier handling a circulating load of 500 percent. The overflow from the classifier goes to a Hum-mer screen, on which 42-mesh stainless-steel cloth is used to avoid the blinding that occurs with steel-wire screen. The classifier overflow contains 71 percent minus 65-mesh material, compared with 47 percent in the ball-mill discharge. The undersize from the Hum-mer goes directly to the lead flotation cells. The oversize kept out is about 86 percent plus 35-mesh and 95 percent plus 48-mesh in size. The lead flotation concentrate contains 93 to 95 percent of the lead in the ore. The middling particles of sphalerite are freed in two regrinding-classification-flotation stages to float the mineral at as coarse sizes as possible.

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is made: "Reprinted from Bureau of Mines Information Circular 7373."

[2] Senior metallurgist, U. S. Bureau of Mines, Rolla, Mo.

[3] Huttl, J. B., Lead-zinc Flotation Problem Solved in Grinding Circuit: Eng. and Min. Jour., vol. 146, No. 5, May 1945, pp. 94-95.

Several writers have reported progress in the separation of lead from copper minerals. The modification[4/] of the Cerro de Pasco Copper Corporation is to thicken the copper-lead bulk concentrate containing a minimum of residual reagents to get the maximum selective depressive action of the sodium dichromate on the galena with the minimum amount of the reagent, and also to make permissible a 4-hour conditioning period.

High-density medium plants fit present trends, in that they have only a few large units and help to simplify the flow sheet for ores containing considerable barren gangue with much of the valuable minerals as fairly coarse aggregates. To lessen grinding as well as crushing costs, with the same recovery of minerals as obtained in the typical gravity concentrator, the high-density medium plant must separate more barren material from feed usually crushed but once than the older gravity plant had to separate from a series of stage-crushed middling products. Plants have been modified to allow treatment of finer particles, and developments may be expected with this new process to permit the separation of middling products for recrushing and retreatment.

The Humphreys spiral classifier[5/] offers promise in the separation of fine sands.

The achievements at Port Pirie may be considered a helpful criterion of what may be done by using larger and more continuously operated smelter units to increase production per man-day and yet improve metallurgy, when climate, ore supply, and other factors together give the most uniform operating conditions enjoyed by any lead smelter (Proc. Aust. Inst. M. & M. since 1930). G. K. Williams[6/] stated 37.9 man-days were required for the production of 1,000 tons of lead in the refinery and 0.623 man-day for 1,000 ounces of refined silver. The labor requirement for other units is likewise stated to be low. There is a steady flow of lead through the softening and following processing units, each adjusted to give maximum reaction rates or optimum solubilities for the reacting components, or temperatures for self-skimming of products, as the metallurgy indicates to be best.

[4/] Wright, T R., Ore Concentration: Min. and Met., vol. 26, No. 467, November 1945, p. 534.

[5/] Humphreys, I. B., and Hubbard, Judson S., Where Spirals Replaced Tables, Flotation Cells: Eng. and Min. Jour., vol. 146, No. 3, March 1945, p. 82.

Gleeson, George W., Why the Humphreys Spiral Works: Eng. and Min. Jour., vol. 146, No. 3, March 1945, p. 85.

[6/] Williams, G. K., Continuous Lead Refining at Port Pirie, South Australia: Trans., A.I.M.E., vol. 121, 1936, p. 226.

The available supplies of lead and the present high capacity of American blast furnaces make it unlikely that furnace sizes will be increased greatly. The increase in width of Dwight and Lloyd sintering-machine pallets from 42 to 63 inches, however, brought very favorable results at three American plants. Others have considered 83-inch wide pallets more necessary to a needed overhauling in material handling, or to operate fewer shifts, than for increasing capacity. The design of flue systems to lessen gas volumes handled and to permit classification of dusts into better-separated products is also being considered. The war system of smelting whenever operators are available from other plant duties has not encouraged semicontinuous lead operations.

Shutting down gold properties removed some of the difficulties in lead smelting caused by arsenic as an impurity. Better drosses, such as those containing 48 percent copper to 10 percent lead, have been made. R. A. Parry[7] found that by using soda ash, 3 to 5 percent of coke, and a little litharge in his rough drossing reverberatory, he could lower his drossing temperatures from about 1,900° to between 1,500° and 1,700° F. and thereby diminish miscibility of the lead and copper. He made copper dross containing 60 percent of copper to 8 to 10 of lead, instead of 45 copper to 18 of lead, and the lead bullion left the reverberatory containing only 0.5 percent copper instead of 2 or 3 percent.

The "fuming" of zinc from molten-lead blast-furnace slags has made it advisable to seek cheap and usually low-grade oxidized zinc products to make more uniform slags and to seek more lead supplies to keep the blast furnaces going. The reduced lead content of sinter charges, also caused, in part, by the inclusion of products reclaimed from tailings and other sources, has helped to lower their sulfur content to around 1.2 percent and to improve the structure of cakes. A little coke breeze added to both "overs" of sinter has improved the product, kept the amount of scrap iron in the blast furnace charge to less than 3.5 percent, and reduced the consumption of coke in the blast furnace to around 9.5 or 10 percent, and even to 8.5 percent when old slags formed part of the charge. Consistent blast-furnace tonnages of 670 tons per day and considerably higher shorter records have been reported.

The latest of the "slag-fuming" furnaces to make zinc oxide was desribed by the Bunker Hill staff,[8] with their own improvements and their suggestion for the future better handling of the deleading of the fume. Methods for obtaining better and quicker contact of reducers in the molten bath are being considered at all plants. The St. Joseph's electric slag-treatment furnace at Herculaneum, Mo., which makes metallic zinc of prime western grade, was described by W. T. Isbell and Carleton C. Long.[9] They forecast that 75 percent of the zinc in the molten slag will be recovered, 68 percent ot it in the "first pass," with "margin to spare." Much of the success of the large electric furnace unit is due to the Weaton-Najarian vacuum condenser. It is suggestive that Mr. Najarian[10] ends his article thereon with the thought of promise for the St. Joseph Lead Co. in the years to come.

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[7] Perry, R. A., Treatment of Speisses and Drosses as Produced in Lead Smelting: Trans., A.I.M.E., vol. 159, 1944, p. 75.

[8] Feddersen, P. C., Schuettenhelm, J. B., Lee, H. E. Gittinger, D. R., and Dunn, G. W., Design and Operation of the Bunker Hill Slag-treatment Plant: Trans., A.I.M.E., vol. 159, 1944, p. 110.

[9] Isbell, W. T., and Long, Carleston C., Direct Production of Metallic Zinc from Lead Blast-furnace Slag: Trans., A.I.M.E., vol., 159, 1944, p. 176.

The large Scotch hearth plant at Alton is well known as a major smelter of lead from rich concentrates. There are, however, other large producers who could put about 80 percent of the lead in their present concentrates into an 80 percent product, if there were incentive to do so. It was shown on an experimental scale[11] that if the lead sulfide was altered to the oxide or to the sulfate, the product could be satisfactorily desulfurized by sintering regardless of the richness of the lead. (There was considerable metallic lead in the bottom of the cake; in practice, provision would be needed for removal of that part of this lead that would enter the windboxes, as at present designed.) A 12-foot mechanized Scotch hearth is now being constructed by the Eagle-Picher at Galena, Kans.

Antimony, bismuth, and other metals besides zinc are being obtained as byproducts from lead ores. The most important unutilized metallurgical byproducts, from the standpoint of our national economy, are those needed for agricultural fertilizers. By circulating the sulfur dioxide gases from the Dwight and Lloyd sintering-machine windboxes, the American Smelting & Refining Co. metallurgists can make 6 percent sulfur dioxide gas out of all the sulfur received. Gas at this concentration supplies enough heat for economical acid making; with little of the sulfur dioxide evolved to the atmosphere the sulfur can be roasted from lead ores without damage to vegetation near centers of fertilizer consumption.

This summary is from the experience of field operators, whose courtesy is gratefully acknowledged.

10/ Najarian, H. K., Weaton-Najarian Vacuum Condenser: Trans., A.I.M.E., vol. 159, 1944, p. 161.

11/ Kuzell, C. R,, Clarkdale Method of Hot-patching Operating Furnaces: Trans. A.I.M.E., vol. 159, 1944, p. 44.